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MILLING METHOD AND COST AT THE CONGLOMERATE MILL OF THE CALUMET & HECLA CONSOLIDATED COPPER CO.

BY

C. HARRY BENEDICT

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MILLING METHOD AND COST AT THE CONGLOMERATE MILL OF THE CALUMET & HECLA CONSOLIDATED COPPER CO.[1]

By C. Harry Benedict[2]

This paper presents the details of milling practice and costs at the conglomerate mill of the Calumet & Hecla Consolidated Copper Co., Lake Linden, Mich., and is one of a series on milling methods being published by the Bureau of Mines.

GENERAL DESCRIPTION

The mines of the Calumet & Hecla Consolidated Copper Co. are located in the Keweenaw Peninsula of northern Michigan. The mines of this district lie in a belt not over 4 miles wide and 100 miles long, which contains more or less commercial ore. The Calumet & Hecla properties are the farthest north, except for the Seneca, of the operating mines. The district borders on Lake Superior, and by means of the Portage and Torch Lake waterways the largest lake vessels can transport coal and other supplies from the lower Lakes directly to the company docks.

The Calumet & Hecla mines are located on a plateau or ridge some 600 feet higher than Lake Superior and its concentrators all border on Torch Lake. The maximum rail haul from the mine does not exceed 10 miles. Water is abundant, and its use at times in the past has been almost too free. Coal transportation is accessible from the Pennsylvania and West Virginia fields, so that isolated steam power and more particularly centrally generated electric power from coal are abundant and inexpensive. Hydroelectric power is being developed in the district, but electricity therefrom is not available at present. Electric power is supplied by two mixed flow turbines of 7,500 and 10,000 Kw., driven by high-pressure steam from the boiler house at 175 pounds and low-pressure exhaust from the stamps at 1 pound above atmospheric pressure. Power is generated at 4,000 volts by 3-phase, 25-cycle generators and stepped up to 13,200 volts for transmission to mines and down again to 2,300 volts for use. The main boiler plant contains twenty-four 500 -hp. Babcock & Wilcox boilers with Roney stokers.

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2 One of the consulting engineers, U.S. Bureau of Mines.

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The mills of this company are all located within 2 miles of each other along Torch Lake. This lake, about 4 miles long and 1 mile wide, is quite deep and is surrounded by gently sloping hills, so that it has been an ideal situation for both the concentrator for current production and the reclamation plant for re-treating old tailings.

There are three groups of mills. At Lake Linden are the original stamps for treating conglomerate rock - some 28 units, each having a daily capacity of 350 tons conglomerate, only 11 of which are now operating. The reclamation plant is also there and has a capacity of 4,000 to 6,000 tons daily, the tonnage depending upon the fineness of the sand treated. The coal docks and smelter adjoin the mills on the south, and south of these is the Ahmeek mill - eight stamp units of 6,500 tons total daily capacity on amygdaloid. Half a mile south of this is another group consisting of the 7-unit Osceola mill, the 4-unit Lake mill, and the Tamarack reclamation plant. The combined capacity of all these units is in excess of 25,000 tons daily of mine rock and 8,000 tons of tailings. The deposits being viewed are the Calumet & Hecla conglomerate, the Osceola lode on the original Calumet & Hecla property, and the Kearsarge lode at Ahmeek, Allouez, and North Kearsarge. The tailings plants are treating the Calumet & Hecla conglomerate at Lake Linden and the Tamarack conglomerate sands deposited at Hubbell.

The conglomerate and amygdaloid ores, although similar in that native copper is the economic mineral in both, are so different in the physical nature of the gangue and in the distribution and size of copper particles that decided variations in metallurgical practice are necessary. The reclamation process for treating tailings, aside from the mechanical means of obtaining the sand and its distribution to the plants, coincides in practice with the treatment of conglomerate sand of the same size. In fact, at Lake Linden, after secondary crushing in pebble mills, the reclamation product and the mine product are combined for treatment by leaching and flotation as there has been no chemical variation in the character of the ore during the 50 odd years that it may have lain in the lake, above or below the water line. The Calumet conglomerate is the richest ore found in the district. The recovery has been as high as 100 pounds to the ton in the early days of the mine and has run consistently over 40 pounds in recent years, with the bottom levels yielding somewhat below this figure.

OCCURRENCE OF THE COPPER

Copper occurs only as native metal, with too little oxide and sulphide to be expressible in percentage. The metallic particles range from microscopic size to masses that require special handling at the mine and are sent to the smelter under the name of "mine mass". This mine mass amounts to about 0.01 per cent of the total copper in the conglomerate, whereas in some amygdaloids (Isle Royale) it is as high as 9 per cent. Moreover, copper particles too coarse to enter the stamps (mill mass) constitute 0.2 per cent of the total copper in the conglomerate as against 10 per cent in Isle Royale amygdaloid.

The conglomerate consists chiefly of felsite, feldspar porphyry, and quartz porphyry, with some few amygdaloid bowlders. In some parts the lode consists largely of coarse to fine sand; in other portions it is a pebble conglomerate.

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The hanging wall is a dense trap rock and the footwall a scoriaceous amygdaloid. The gangue minerals are calcite, epidote, and feldspar, with some barite and chlorite.

The copper occurs chiefly as cementing material around the pebbles. There is some very fine copper within the pebbles, and the sand contains copper of microscopic fineness. Complete liberation of values is therefore impossible. Leaching 100-mesh ore yields recoveries of more than 90 per cent, but flotation attains this recovery only with 200-mesh ore. Accordingly leaching is indicated for the crystalline sands and flotation for the slimes.

The ore has a hardness of 7 and is very tough. There are no fractures once the conglomerate is broken down. Comparative tests on resistance to crushing are difficult to obtain, but such independent work as has been done by outsiders indicates that the Calumet conglomerate resists crushing and particularly fine grinding more than any other known copper ore.

The ore has a high abrasive action corresponding to the resistance to crushing as evidenced by the great wear of stamp shoes in stamping, of pebbles or balls in grinding, wearing parts in pumping, etc. The chilled iron shoes stamping conglomerate have only one-tenth the life of shoes stamping Osceola amygdaloid.

COARSE CRUSHING

The rock as broken underground is dumped on a grizzly with about 4-inch openings when new, and the oversize is crushed in Blake jaw crushers to the same size. The breakers receive material as large as can be conveniently handled in tram cars and skips but not larger than about 24 by 30 by 36 inches. A locally designed crusher of the Blake type, 24 by 48 inches, is standard practice. The crushers are driven by an endless belt from a motor-driven countershaft, and each has a capacity of about 200 tons per hour.

The percentage of fines is too small to warrant any preliminary screening out before stamping, as has been amply proved by several unsuccessful installations with this refinement in mind. The largest rock that can be fed to the stamp without choking the feed hoppers is 10 by 12 inches, and primary crushing in the concentrator begins at this size. The stamp discharge screens have 3/16-inch round openings.

PREVIOUS HISTORY OF CONCENTRATOR OPERATIONS

Lake Superior copper-concentrator practice differs from that of any other district owing primarily to two unique characteristics of the ore: (1) Native copper is present in all sizes from masses weighing many tons to microscopic size, which resists reduction in size when once liberated except by abrasion, and (2) the particles of native copper are tougher than the ore and hence are more resistant to sliming.

Careful analysis of these two properties of the ore accounts for much of what has been considered by some metallurgists as inefficient metallurgical practice and explains why the steam stamp has reigned supreme in the Lake copper district for 50 years in spite of an occasional attempt to replace it, although it has made no headway in other camps. It explains why stage crushing has not proved successful against single reduction from 10-inch through 3/16-inch mesh

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at one operation and the prevalence of jigs and roughing tables in spite of their recent abandonment in favor of all flotation in many camps. A piece of copper the size of the head, the fist, or even of a pea presents rather a firm argument against either graded crushing or all sliming practice.

Stamps were used on Lake Superior in the sixties, as soon as the mines developed any tonnage. Mechanically, the efficiency was low and was so recognized. The compound stamp (high and low-pressure cylinders), developed about 1900, was a considerable improvement on the simple stamp; but modern practice is to run the stamp, simple or compound, noncondensing, and to use the exhaust steam in low-pressure turbines, thereby obtaining an excellent mechanical efficiency from the combined unit.

The only serious effort to displace the stamp as a primary crusher was made some 20 years ago by the Copper Range Co., which attempted stage crushing by means of rolls with intermediate jigging. It gave no promise and was soon abandoned. A Symons cone crusher might be able to master the nuggets of copper that clog other types of crushers or rolls, but existing mines are fully equipped, and with low stamping costs there is little inducement to warrant the capital expenditure necessary for a change in method.

Current practice calls for hand picking of large nuggets from the rock, preceding stamping. When a large piece of copper escapes the vigilance of the picker and goes into the mortar it does no harm. It remains there, abrading slowly, until such time as a shoe is changed, when the content of the mortar is shoveled out if necessary and treated in a specially designed jig. Standard practice up to about 1900 was to clean out the mortar periodically, as often as every four hours in the early period of rich yields from the Calumet conglomerate, and to wash out the large nuggets of copper. About 1900 two devices were invented to remove this copper from the mortar hydraulically. The Krause mortar discharge was an hydraulic discharge of the ordinary type large anough to remove 4-inch nuggets, the copper within the mortar working down against the water pressure and being trapped off periodically. The Woodbury mortar jig accomplishes the same purpose by means of a pulsating current operating within the mortar at the stave line and is effective not only for large pieces of copper but for smaller nuggets down to the 3/16-inch size that will pass through the mortar screen. This class of material constitutes only 3 per cent of the total copper recovered from the conglomerate but as high as 30 per cent of that from rich amygdaloid.

Before the introduction of the Wilfley table in 1898 the recovery of copper from slimes (local term for minus 60-mesh material) was very inefficient. Fortunately the values in the slimes were low because of the nonfriable nature of the valuable mineral. Revolving tables were used on primary slimes only, as the recovery of fines was so low that fine grinding by rolls or Chilean mills was not economic. With the gradual displacement of the revolving tables by Wilfleys, finer grinding was introduced, first by means of Chilean mills and later by Hardinge pebble mills. These pebble mills are still the standard fine-grinding units for treating conglomerate, ball mills being much more costly to run than pebble mills because of excessive abrasion of iron due to the hard rock. Leaching of fine sands was begun in 1916 and flotation of slimes soon afterward, so that the practice has not changed fundamentally for the past 12 years, and there

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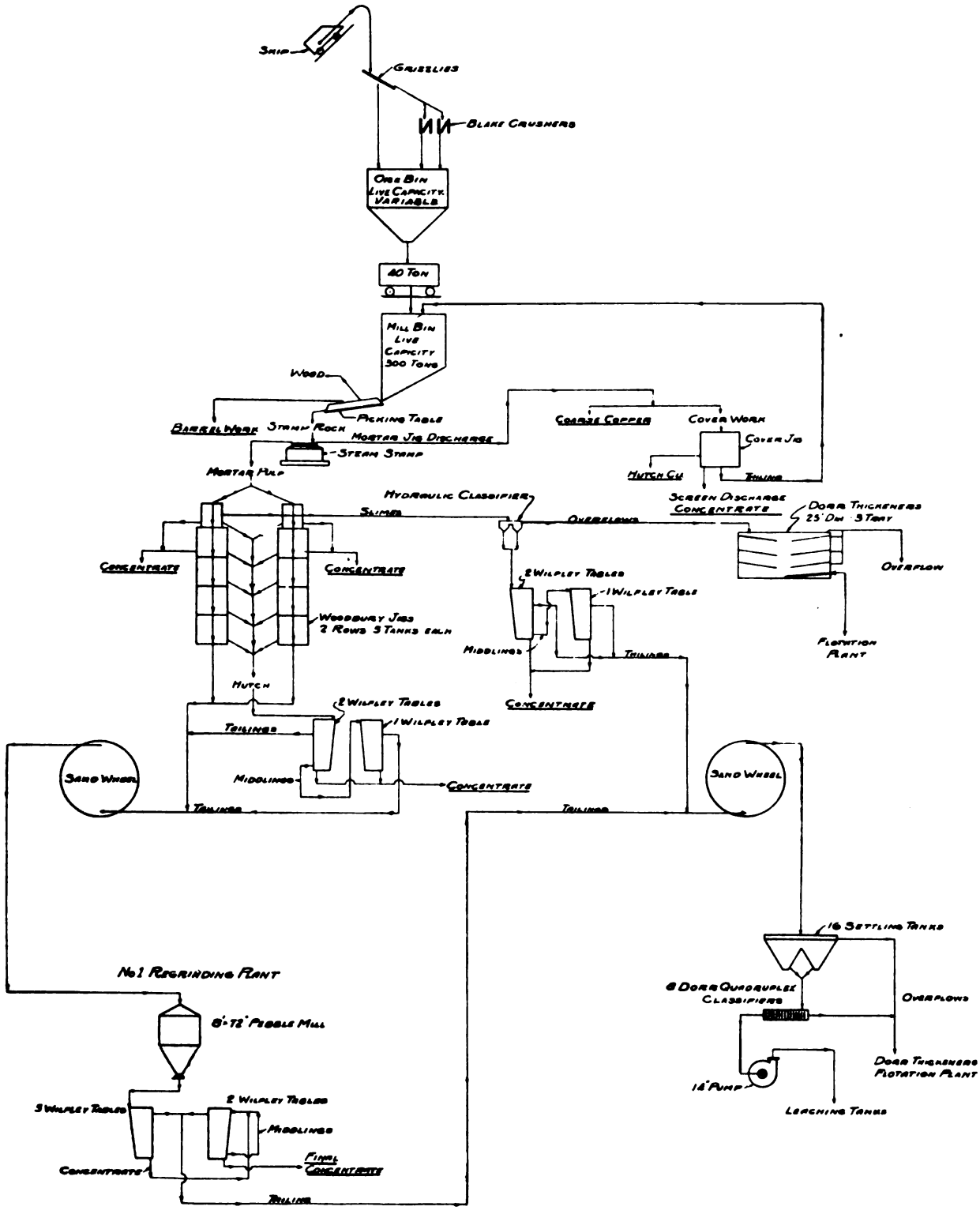

Figure 1.- Gravity flow sheet of Calumet and Hecla conglomerate. One unit Calumet mill; one unit regrinding plant; sand and slime classification

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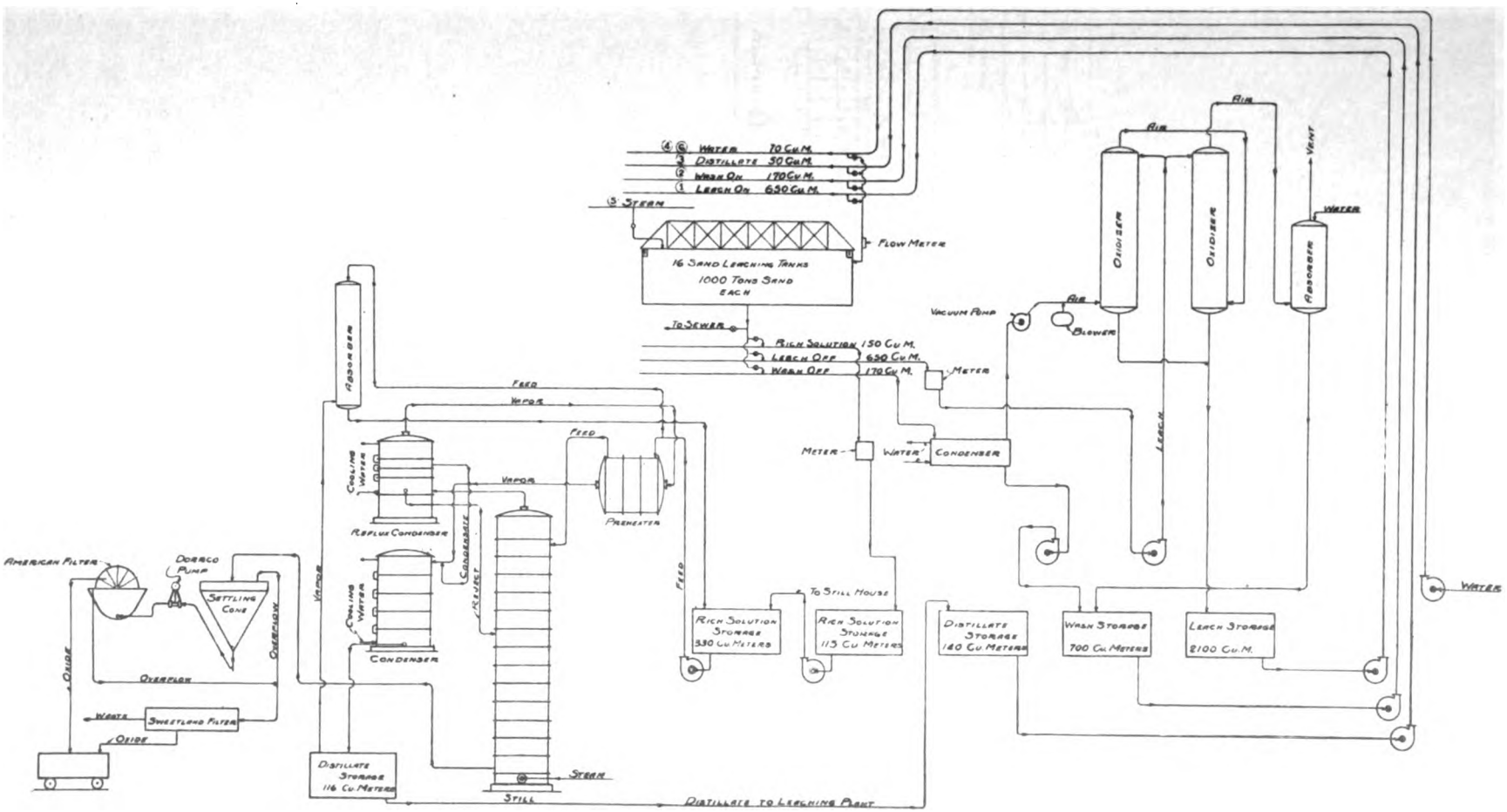

Figure 2.- Flow sheet of ammonia leaching

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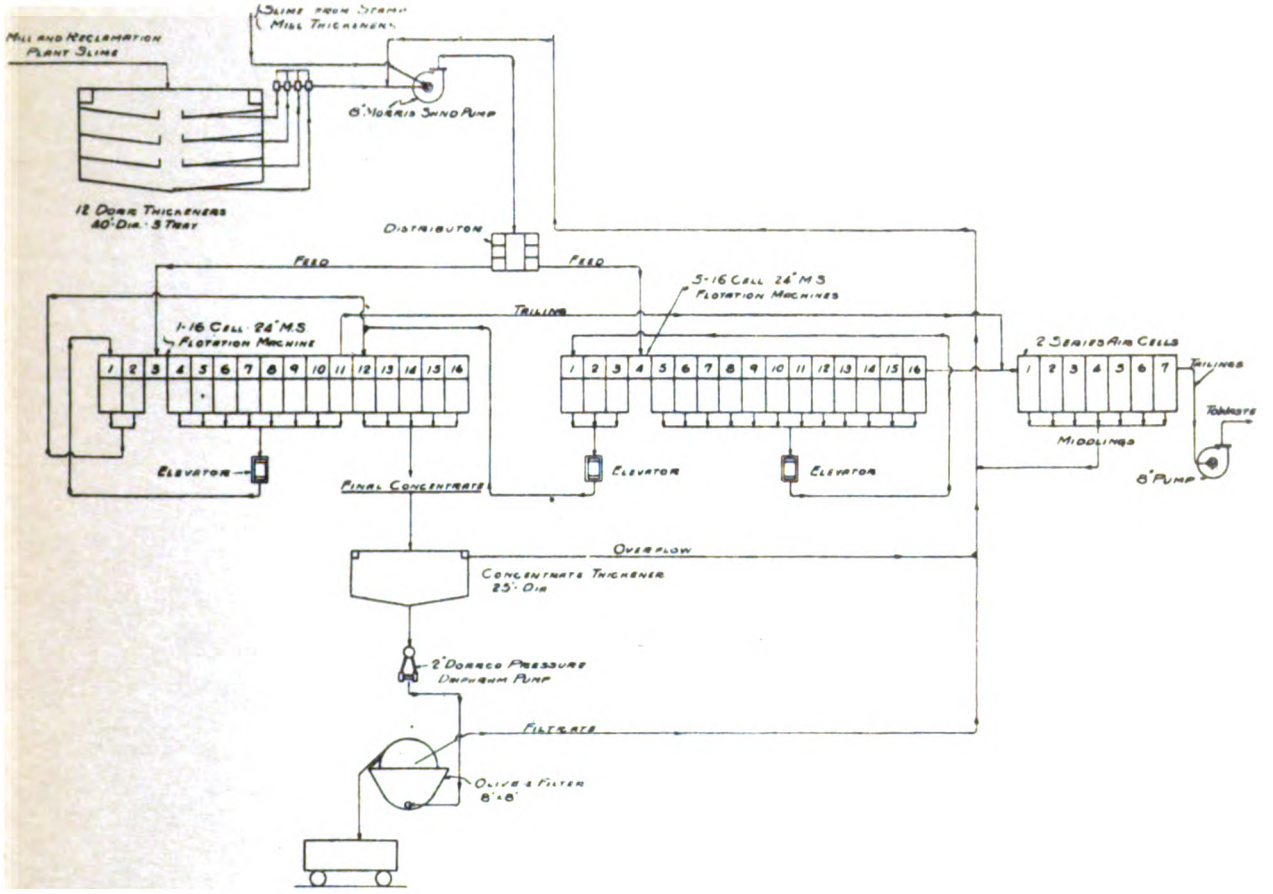

Figure 3.- Flotation flow sheet of Calumet and Hecla conglomerate

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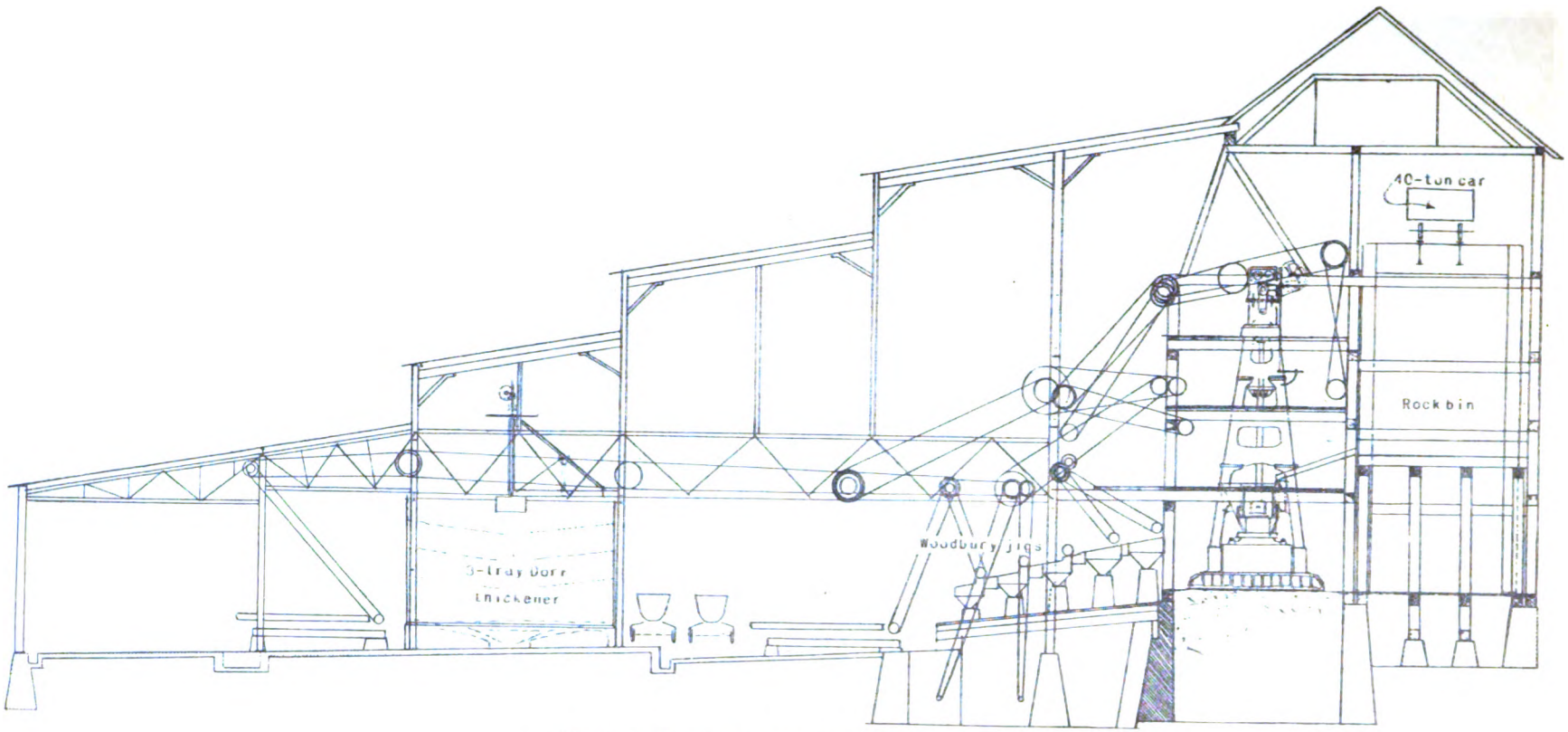

Figure 4.- Sectional elevation of Calumet stamp mill

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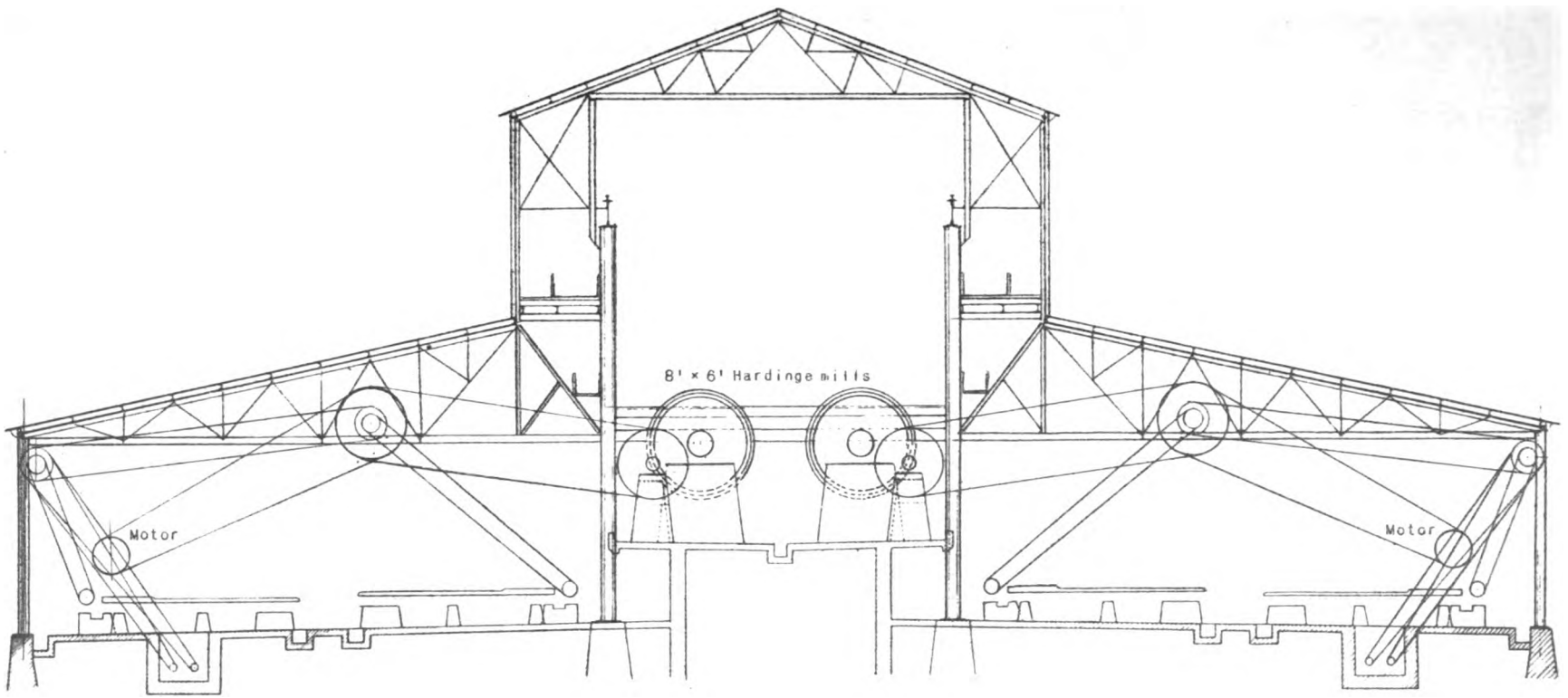

Figure 5.- Sectional elevation of No. 1 regrinding plant

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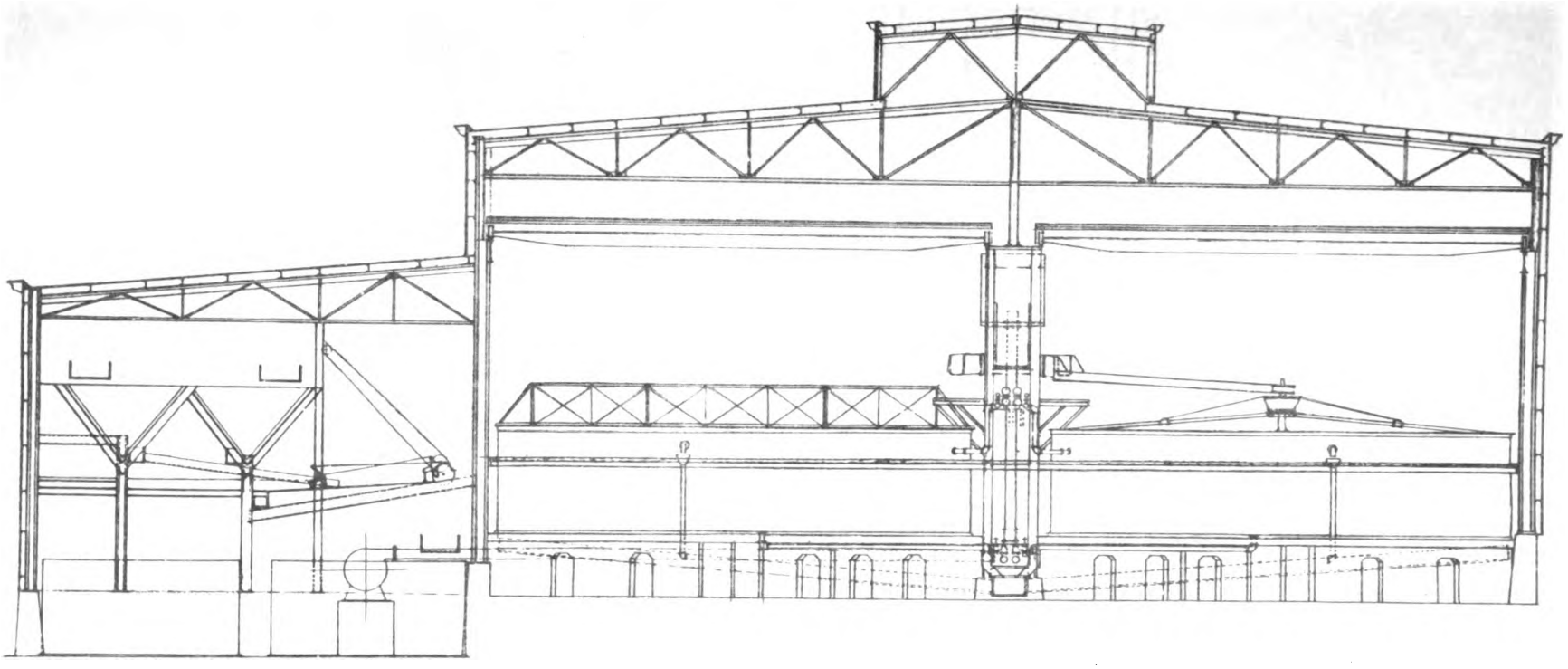

Figure 6.- Sectional elevation of leaching plant

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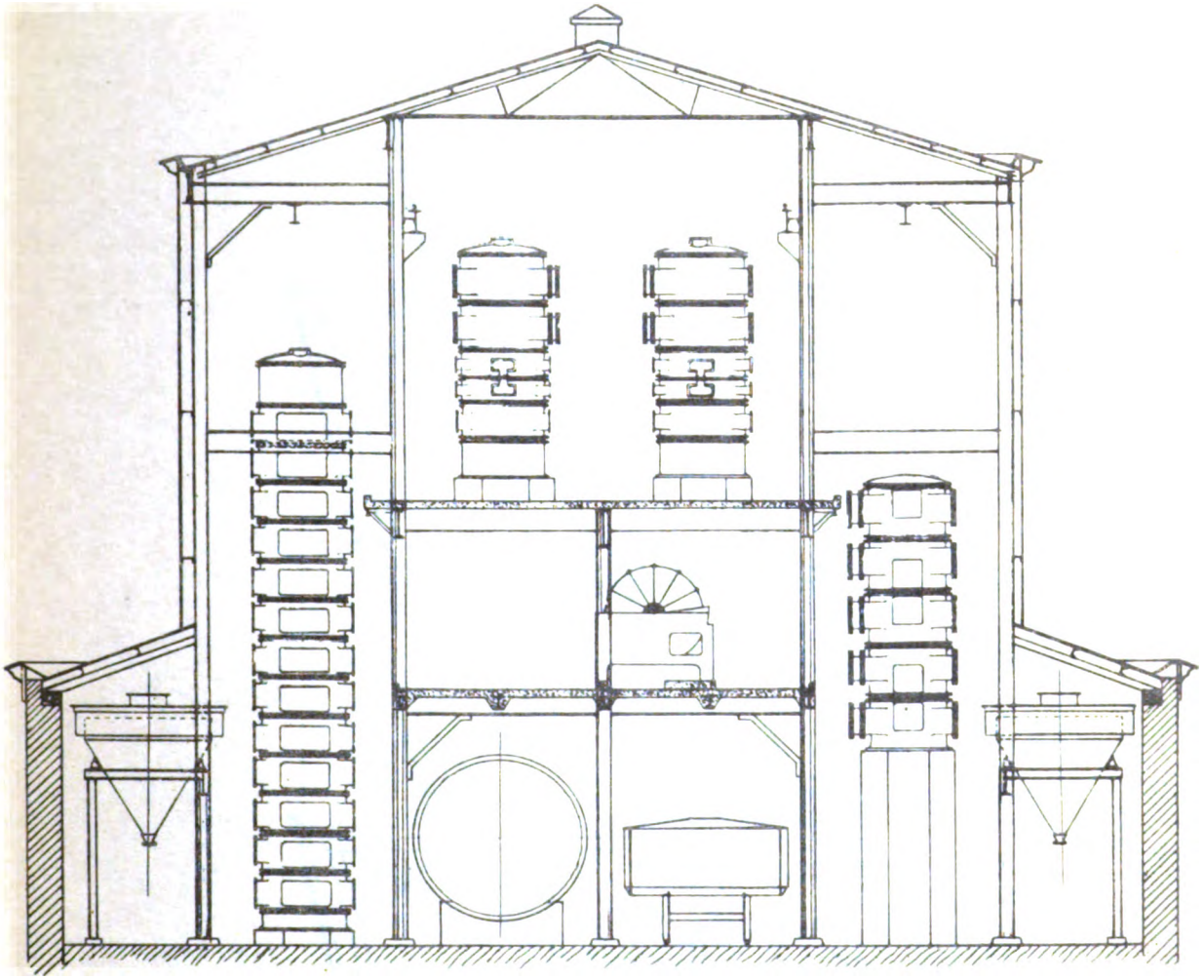

Figure 7.- Sectional elevation of distillation plant

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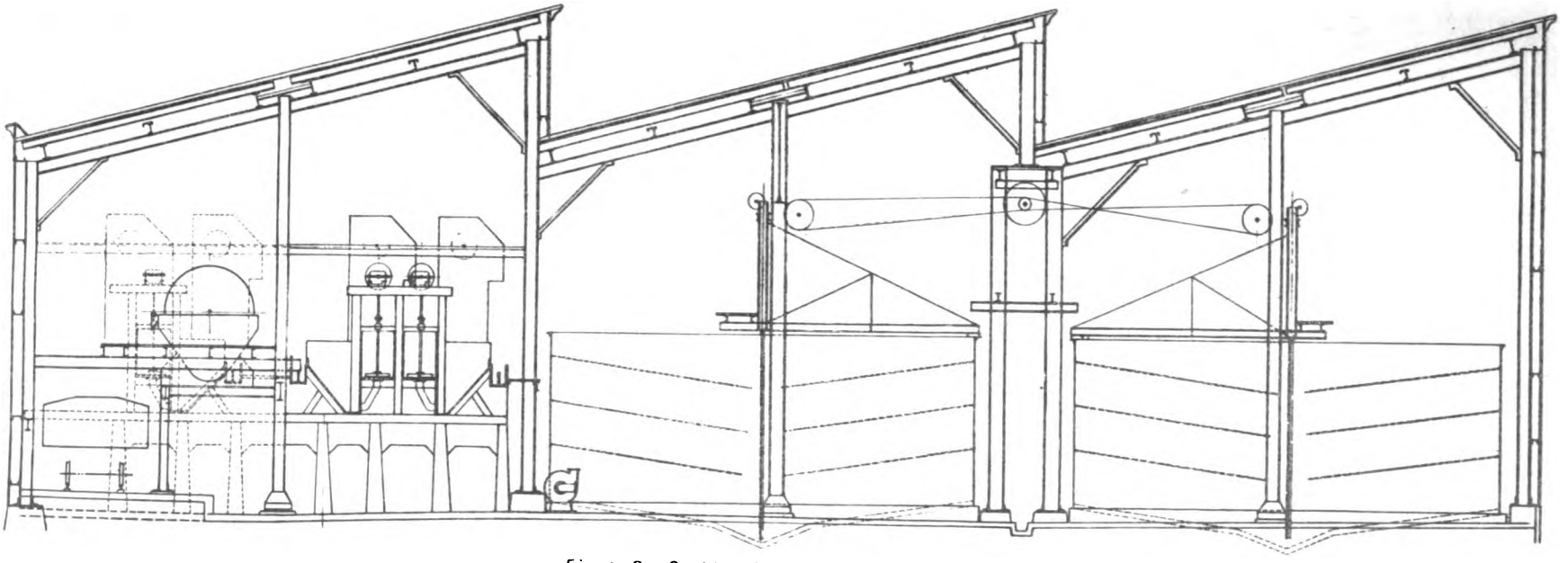

Figure 8.- Sectional elevation of north flotation plant

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have been no notable additions to existing plants and facilities in that time. The metallurgical efficiency, however, has been improving from year to year, as the following table shows:

Year	Assay Concentrates, per cent	Assay Tails, per cent	Assay Heads, per cent	Extraction, per cent
1900	61.11	0.783	3.423	77.92
1905	61.96	0.667	3.203	79.94
1910	56.79	0.450	1.956	77.46
1915	58.50	0.463	1.950	76.72
1920	57.98	0.179	1.934	91.01
1925	63.17	0.143	2.246	93.83
1926	62.18	0.139	2.260	94.49
1927	60.18	0.117	2.187	94.61
1928	61.31	0.114	2.260	95.12
1929	61.27	0.124	2.223	94.62

PRESENT METHOD OF CONCENTRATING

The concentrator as at present operated consists of four main units, separately housed, as follows:

1. Calumet stamp mill.
2. No. 1 regrinding plant.
3. Leaching plant (including still house).
4. Flotation plant.

Figure 1 is a flow sheet of the gravity concentration as exemplified by one unit of the Calumet mill (of which there are 11), one unit of No. 1 regrinding plant (of which there are 24), and complete sand and slime classification. Figure 2 is a flow sheet of the leaching plant and still house, and Figure 3 is a flow sheet of the flotation plant. These last two plants treat not only current mine product but also reclaimed sands. A sectional elevation of the Calumet stamp mill is shown in Figure 4, a corresponding elevation of No. 1 regrinding plant in Figure 5, of the leaching plant in Figure 6, of the still house in Figure 7, and of the flotation plant in Figure 8.

Intermediate crushing is entirely by Leavitt stamps. These have single cylinders with upper inlets and exhaust ports only, the upper piston working against a piston in the lower cylinder without ports, which acts as a steam spring. The advantage of this type of simple stamp is that there is no clearance loss at the bottom of the stroke. There is, however, a cushioning effect to the blow, that limits its effectiveness.

The stroke is 24 inches, the weight of shoe 730 pounds, the weight of moving parts 6,000 pounds with shoe and shaft new, the mean effective pressure 80 pounds, and the horsepower 195.

The stamp screens have 3/16-inch round openings punched in a plate 0.11 inch thick. A typical screen sizing of mortar pulp is as follows:

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Size		Per cent	Cumulative per cent	Assay, per cent
Plus	10 mesh	14.5	14.5	5.61
do	20 do	14.4	28.9	4.28
do	28 do	8.3	37.2	4.34
do	48 do	13.4	50.6	3.34
do	65 do	6.1	56.7	2.23
do	100 do	6.2	62.9	1.91
do	150 do	3.6	66.5	1.91
do	200 do	5.2	71.7	1.58
Minus	200 do	28.3	100.0	1.09
	Total	100.0		2.95

Coarse copper is discharged by means of a Woodbury mortar jig, which makes two products, a screen discharge which may have nuggets 1½ inches or more in smallest diameter and a hutch product all of which has passed through the 7/8 by 2½-inch openings of the screen. This jig is actuated by a plunger outside the mortar. The opening into the jig chamber is at the line of the base of the mortar screens. There are two pockets, one for the hutch product, which fills continuously, and one for the oversize, which is discharged intermittently.

PRIMARY GRINDING

Jig tailings and finisher tailings, material all minus 3/16-inch and approximately plus 20 mesh, is ground to a maximum of minus 35 mesh in Hardinge mills. These mills, 8 by 6 feet, are run in open circuit at a speed of 24 r.p.m., and are belt-driven in units of three from a 250-hp. motor. There are 24 of these mills in a separate plant (No. 1 regrinding plant), only about half of them being operated at present.

Each mill has a capacity of about 110 tons per 24 hours, requires 90 horse power, and is run with about 25 per cent of solids. The mills are lined with old steel rails, 60 pounds or heavier, imbedded in neat cement. Danish pebbles, preferably of 4-inch size, are used. The pebble consumption is about 5 pounds per ton of sand ground, and the pebbles cost about 3/4 cent per pound delivered or 4 cents per ton of sand ground. Steel balls have been tried and abandoned only because of excessive cost, as the wear per ton of sand ground is about 80 per cent of that of pebbles. Liners last about 11 months, during which time 30,000 to 35,000 tons of sand is ground.

The cost of primary grinding per ton for 1929 is as follows:

General expense		$ 1.85
Conveying & distribution		.38
Grinding		12.64
Attendance	1.01	
Power	7.78	
Pebbles	3.28	
Lining	.57	
Washing		2.13
Total		$17.00

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SCREENING AND CLASSIFICATION

For classifying out the original slime after sampling, a Woodbury jig classifier is used. This machine makes four products: (1) Slimes at about 60 mesh, (2) hutch product, (3) coarse copper concentrates, and (4) coarse and fine sand. This Woodbury classifier was developed at the Calumet & Hecla and is generally used in the Lake copper district but has not made much progress in other fields. It is now being displaced to a great extent by the Dorr classifier, which makes only two products whereas the Woodbury classifier effects considerable concentration of the copper values, acting as an ordinary jig in this respect, and also removes the slimes.

The slimes are removed by means of two brass shields or cups at the tail of the sieve. The outer cup, 8 inches wide and 12 inches long, is attached to the tail board of the jig and is so set that its bottom level is in the sand, thus effectively shielding it from entrance of slimes that pass over the tail board at the sides of the shield. Within this shield is placed the copper discharge cup, its lower edge extending into the moving bed of copper concentrates. It is thus sealed against the entrance of sand which discharges through the tail board between the two shields while the copper itself is drawn off from within the inner shield.

The Woodbury classifiers have 24 by 36 inch screens with plungers 12 by 24 inches, and operate ordinarily at 195 r.p.m. There are two such classifiers per stamp unit. Mortar water amounts to about 350,000 gallons per 24 hours, which with 350 tons of ore gives a slime density of about 1:4 by weight. The slimes are further classified by a hydraulic classifier of ordinary launder type, and the spigot product is subsequently treated on Wilfley tables to save the coarsest of the fine copper (60 to 200 mesh). The classifier overflow, about 93 per cent of which is less than 200 mesh, is thickened in Dorr thickeners and treated by flotation.

Launders within the mill and outside are of wood with steel plate liners. Rubber has been used experimentally and gives promise of being more economical than steel on coarse sands where abrasion is severe. For fine material, where steel lining lasts for years, the first cost of rubber lining is prohibitive. The slope of launders ranges from 1-1/2 inches per foot on 3/16-inch maximum particle size in mortar pulp with a water ratio of 4:1 to 3/16-inch per foot with a water ratio of 20:1 and higher.

HAND SORTING

Hand picking is done only before stamping, more because of mechanical than metallurgical considerations. Ore from the rock bins is fed to the stamps by means of a shaking apron, and the tonnage is controlled by an attendant whose chief duty is to regulate the feed so that the stamp strikes its full blow of 24 inches. A shorter blow spells mechanical inefficiency, whereas a longer one endangers breakage of the cylinder heads or piston. The attendant picks out wood and other rubbish and large masses of copper that would lodge in the mortar or jam between the shoe and die. This copper is called "barrel work," a local term handed down from 30 years ago when it was packed in special barrels for shipment to the smelter.

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GRAVITY CONCENTRATION

The original mills, the first units of which were built in 1868, still constitute a portion of the present plant. The rock-bin columns, 14 by 14 inches, built of white pine, still support the load of rock although steel columns now carry the weight of the rock cars. Steam stamps have been made heavier and more powerful since the days of the Ball heads. The first Leavitt stamp was erected in 1879, but revolutionary changes in this department in the last 50 years have been few. There has been a gradual increase in steam pressure with a corresponding increase in stamp capacity, and steam economy has been effected by the use of a low-pressure turbine for the exhaust; however, the original stamps are still pounding away.

The concentrating building and machinery, the latter consisting chiefly of Woodbury jigs and Wilfley tables, dates from 1903. In 1907 the No. 1 regrinding plant was erected for finer grinding and tabling of jig and coarse table tailings. It was equipped originally with 48 Chilean mills, 6 feet in diameter, but these have since been replaced by 24 Hardinge pebble mills, 8 feet in diameter by 6 feet long. This equipment marked the limits of recovery by gravity concentration; and it was not until leaching was introduced in 1915 and flotation in 1917 that recoveries could be called satisfactory, although they had previously fully equalled the recoveries of corresponding concentrating plants in the copper industry.

Woodbury jigs and Wilfley tables are practically the only machines used for gravity concentration. Jigs make copper concentrates (locally called "mineral"), hutch concentrates (which are re-treated on Wilfley finishing tables), and tailings. Before the regrinding plant was erected middlings were drawn off for finer grinding but now all of the coarse tailings are reground. Slimes are settled in 25-foot, 3-tray, Dorr thickeners, and the thickened product is drawn off and treated in a separate building (flotation plant) with secondary slimes and reclaimed tailings.

No sampling of heads is attempted, as it is impossible to obtain a representative sample of any size practical to treat and reduce because of the presence of coarse copper. Concentrates and final tailings, both coarse and fine, are sampled daily, but no intermediate products are sampled. Concentrates are graded, chiefly by size, as follows:

Grade	Assay, per cent	Per cent of total
No. 0 Copper (5/8 inch)	91.50	3.69
No. 1 Grade (+10 mesh)	89.36	40.68
No. 2 Grade (-10 mesh)	56.95	55.63

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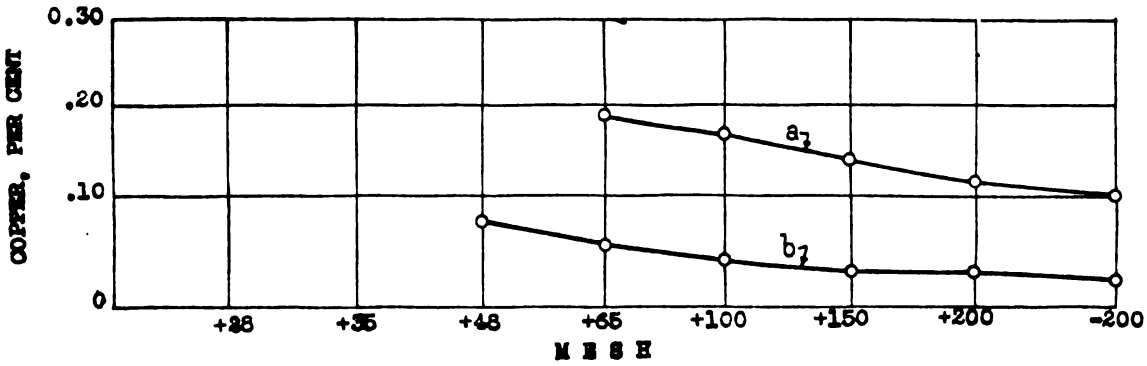

Figure 9.- Assays of screened products from flotation tailings: a, conglomerate tailings; b, amygdaloid tailings

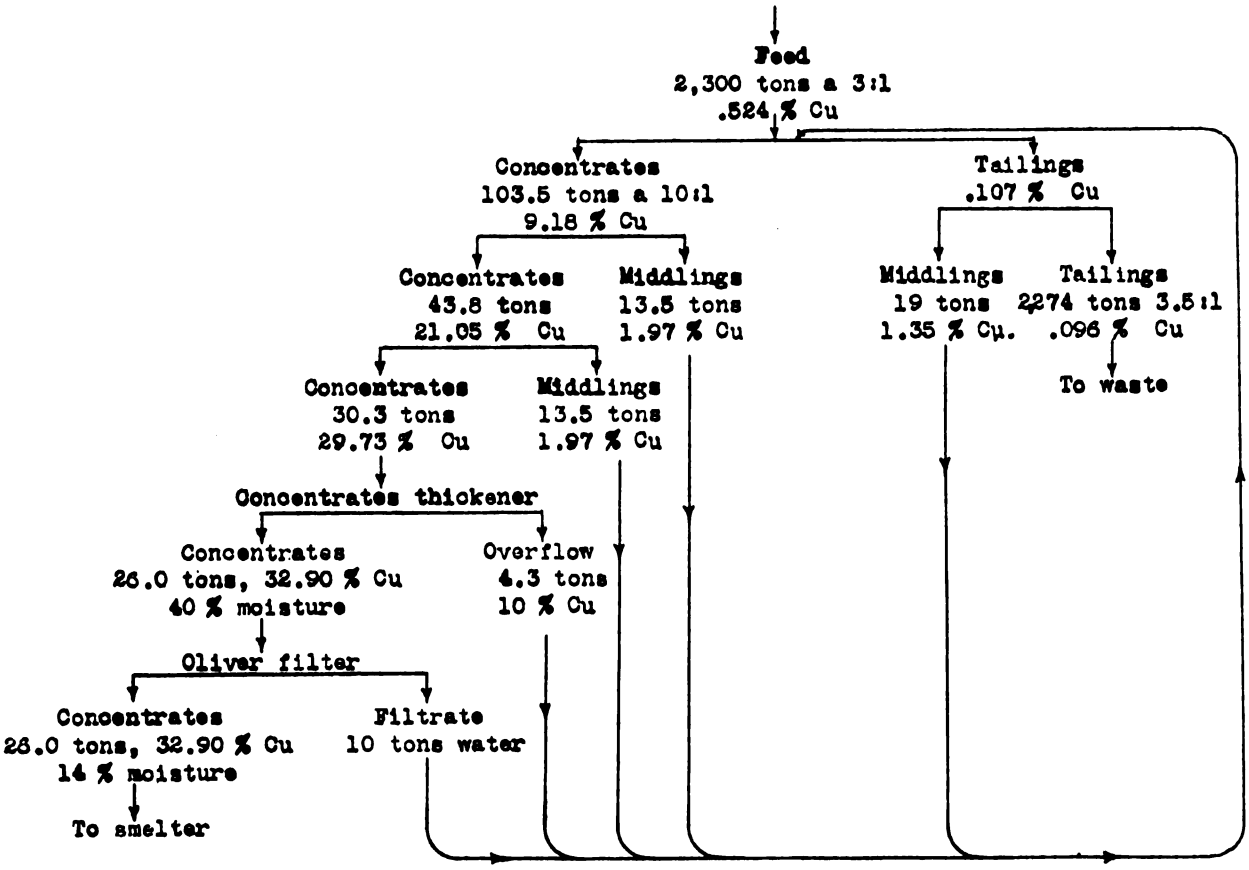

Figure 10.- Flotation circuit

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These figures are for mill concentrates only. Including leaching and flotation recoveries, figures are as follows:

Grade	Assay, per cent	Per cent of total
No. 0 grade (5/8 inch)	91.50	2.79
No. 1 grade (+10 mesh)	86.36	30.82
No. 2 grade (-10 mesh)	54.55	47.78
Flotation (-100 mesh)	35.11	9.36
Leaching (oxide)	81.71	9.25
	61.27	100.00

Following gravity concentration the fine sands (+200 mesh approximately) are leached and the colloidal slimes are floated.

This separation of sands and slimes is effected by means of quadruplex Dorr classifiers, with capacities of 1,000 tons each. The finer tailings are first settled in a series of V tanks. The plug discharge of the V tanks goes to the Dorr classifiers, the overflow joining the overflow of the V tanks for final settling in Dorr tray thickeners.

FLOTATION

The difference in the distribution of the copper values in the amygdaloid and the conglomerate is most marked in the behavior of the ores in flotation. Both kinds of ore contain much coarse copper, but the conglomerate rock also carries a considerable amount of such finely disseminated copper that grinding to minus 200 mesh does not release it very satisfactorily while with the amygdaloids crushing through 48 mesh is sufficient. Graphs made by plotting assays of the screened products from samples of flotation tailings from the two ores show this very clearly (fig. 9). As a consequence only the slimes from the conglomerate are treated by flotation; but with the amygdaloid everything finer than 35 mesh is so treated, although the coarse sand is subsequently tabled to remove the coarser particles of copper that are too heavy to float.

The North flotation plant, which is the oldest operating flotation unit, was put in commission in 1919. The feed consists of thickened primary slimes from the stamp mill and the overflow of the settling tanks and Dorr classifiers from the secondary classifying system. The latter material comes from the regrinding plants handling current tailings and reclaimed sand and from the original fines recovered by the dredge. The quantity treated ranges from 2,000 to 2,500 tons per day, consisting of 1,000 tons of primary slimes, 1,000 tons of reground sand, and a variable amount of fines from Torch Lake.

The slimes from the leaching plant are very dilute, the ratio ordinarily being 50:1 or 60:1 of water to solids, and are thickened to a 3:1 ratio in twelve 4-compartment Dorr thickeners 40 feet in diameter. Eight of these are of the open type, and four are of the connected type. The latter are more efficient and have greater raking capacity than the former. On eight of the tanks the underflow is handled by means of 4-inch quadruplex diaphragm pumps. The combined feed runs to an 8-inch Morris sand pump which elevates it to a distributor above the flotation machines where it is split between them. The xanthate, lime, and part of the pine oil are added to the feed pump to get additional mixing and to avoid the use of individual feeders on each flotation

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machine. About one-third of the pine oil is added to the flotation machines later by means of drip cans. By allowing each can to stand 24 hours after filling and before being put in use no trouble is experienced from choking off.

Clean high-grade lime is used. This lime is slacked in a large volume of boiling water, diluted to a 5 per cent solution, and fed to the feed pump by means of a small diaphragm pump, the rate of feed being regulated by adjusting the length of stroke and speed. The mill water and the ore itself are slightly alkaline, the pH value usually running around 8. A very small amount of lime, however, improves the results. The oil used is a mixture of 75 per cent No. 5 G.N.S. steam-distilled pine oil and 25 per cent A.T.& T. No. 11 destructively distilled pine oil, the latter being used as a froth stiffener. The reagent consumption is as follows:

	Pound per ton
Sodium xanthate	0.05
Lime	0.17
Pine oil	0.15

It is not practicable to use the mill water over again, so the oil consumption is higher than in many western plants where that practice is followed.

The plant contains six 24-inch standard 16-cell Minerals Separation flotation machines of the low-level type, of which four are ordinarily in use. A subaeration machine of the Hebbard type was originally tried out in parallel with a standard machine but did not do as good work. When coal-tar oils were used the tailings averaged about a pound higher than after xanthate was substituted. To cut down this loss two sets of air cells of the Inspiration type, one to each three of the M.S. machines, were installed as scavengers on the standard machine tailings. The additional saving reduced the copper content of the tailings about .02 per cent, but after xanthate was used this additional saving was halved. The concentrates of the air-cell machines runs about 3 per cent copper so they are returned to the feed pump. Air at 4½ pounds pressure is furnished by a No. 1 Connersville blower.

The flotation circuit is shown in Figure 10.

The feed to the flotation machines enters the fourth cell, which is used as a mixing cell only. The last 12 cells act as roughers, the concentrates being returned to the first cell by means of a bucket elevator. The first three cells make intermediate concentrates and a middling which goes back through the roughers. Originally these concentrates were the final concentrates, but as it was found advisable to raise the grade an additional cleaning was adopted. To avoid elevating the froth again the last five cells of the M.S. machine nearest the concentrates thickener were blanked off from the rest of the machine and used for the final cleaning. The concentrates thickener gives a dirty overflow, which is low enough in grade to raise the final tailings of one section when added to the feed to that section but which could not be sent to the concentrate cells directly as it gave too much dirty froth, so it is now run to the feed pump. The product of the concentrates thickener is elevated to an 8 by 8 foot Oliver filter by means of a 2-inch diaphragm pressure pump. Ordinarily the filter cake carries about 14 per cent of moisture. It drops by gravity through a chute into 50 ton hopper-bottom cars and is shipped direct to the smelter.

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The average results for the year 1929 are as follows:

	Per cent	
Assay feed	0.580	
Assay tailings	0.109	
Assay concentrates	35.05	
Recovery	81.2	
General expense		$.0133
Conveying and settling		.0199
Flotation		.0433
Labor	$ .0074	
Oil and chemicals	.0207	
Power	.0123	
Other	.0029	
Royalty		1.00
Total cost per ton treated		$ .0865
Total cost per pound of copper produced		.0100

LEACHING

The leaching plant was operated first in 1916. It has been described in detail in the technical press[3] and the present account will be a brief resume of these articles with developments since they were written.

There have been few changes in the mechanical construction of the leaching plant proper since it was built, although the present cycle of operations bears little resemblance to the original. However, the distillation system for recovery of copper oxide from dissolved solution has been subject to rather decided mechanical changes. Although the plant is almost 15 years old there has been little corrosion in tanks or pipe lines and maintenance costs have been at a minimum. There has been a gradual decrease in costs and an increase in efficiency from year to year, and the operators have been particularly fortunate in that the cost of ammonia and of steam, the two major supply items, have shown and are showing a tendency to decrease.

The plant as at present constituted consists primarily of 16 steel sand-leaching tanks 54 feet in diameter by 12 feet high in two rows of eight each and 25 solution storage tanks with the necessary pumps and piping for circulating solutions. In addition to these there are auxiliary devices such as oxidizing towers, condensing units, etc., which will be described later. This apparatus is all in the leaching plant proper and has to do only with dissolving the copper from the sand. In a separate building, known as the still house, is the apparatus for the distillation of the saturated solutions which consists at present of three units.

The sand for the leaching tanks is made up of mine tailings and reclamation plant tailings that have been classified out by Dorr classifiers. All copper that can possibly be saved by gravity concentration has been extracted, and the remaining copper is chiefly attached to or included in the grains of sand. All sand is supposed to be finer than 28 mesh and, as the sizing assays will show,

3- Eng. and Min. Jour. July 14, 1917. Trans. A.I.M.M.E., February 1924.

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should be finer than 60 mesh for efficient extraction. The sand must be free from colloids. The average sizing will show 15 to 20 per cent of minus 200-mesh material of which only a small portion is colloidal. The tanks, while filling, overflow, and the major portion of the colloidal material remaining after classification escapes from the main body of sand in this overflow.

The solvent employed is supric ammonium carbonate. After distillation the return to the leaching cycle consists of ammonium carbonate, but losses of reagent during the process are made up by aqua ammonia which is carbonated in the plant in the same towers in which the solution is oxidized from the cuprous to the cupric state.

The chemistry of the process is simple. As native copper is the only valuable mineral present in the ore advantage is taken of the two valences of copper in water solution to effect its oxidization to the cuprous form. The first reaction then is between native copper and cupric ammonium carbonate in an excess of ammonium carbonate to yield cuprous ammonium carbonate, weight for weight of copper. To use cuprous ammonium carbonate as a new leaching solution it must be regenerated to the cupric condition by oxidizing it with air in an ordinary oxidizing tower. This reaction between cuprous ammonium carbonate and oxygen takes place in the presence of ammonium carbonate and yields cupric ammonium carbonate which is now ready for the next leaching cycle.

As the loss in chemical solvent is made up by aqua ammonia the solutions would gradually become deficient in carbon dioxide, and this reagent is provided by burning coke in an ordinary Holland furnace and leading the exhaust gas through the oxidizing tower mentioned above. The carbonating and oxidizing is thus done at the same time, and excess of air does no harm. It is only within the last few years that aqua ammonia rather than carbonated ammonia liquor has been used to make up the loss in solvent, and its use has resulted in a very decided decrease in the cost of the ammonia.

After the leaching solution is partly or entirely saturated with copper by passing it through the ore, an amount equivalent to the copper dissolved from the sand is removed from the leaching cycle and is distilled to yield copper oxide with the recovery of the ammonia and carbon dioxide as ammonium carbonate. This leaching cycle theoretically results in no loss of solvent, the only actual chemical used being the oxygen of the air which appears in the copper oxide in the final product. One might trace this product still further to the smelting works where the copper oxide is reduced to the refined metal with the resulting liberation of the oxygen as carbon dioxide, thus partially restoring nature's balance.

The leaching tanks are in two rows. A crane travels over each row of tanks for handling removable tank covers, sand distributors, sand levelers, flushers, and other auxiliaries. As the tanks must be covered during leaching to prevent loss of ammonia by volatilization and to make it possible to live in the plant, all distributing and flushing apparatus must be removable instead of fixed to the building structure as it might be if open tanks could be used. The tank covers are of steel plate with top trusses which are supported on an encircling angle iron around the tank wall. A seal between the cover and the tank is made by a channel iron on the cover projecting into an annular launder on the tank, which acts as an overflow during the sand-filling stage and which is filled with

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water during the leaching stage. This seal prevents the escape of ammonia around the edges of the cover and is the only joint necessary either during the leaching or the steam wash.

For filling the tanks a portable distributor of the Butters and Mein type is used. It is supported on the central shaft of the tank and revolves by its own reactance. The sand is carried throughout the length of the plant in a central launder, and connections are made from this central launder to the distributor by means of a portable launder. Gates in the central launder control the flow so that the sand is led consecutively from tank to tank. The alternating tanks are on either side of the main central launder, and with two sets of portable launders and distributors there is no delay in filling while shifting from a full tank to an empty one.

On the side of the tank at the bottom are six gates and on the bottom is one central cone gate, the opening and closing shaft of which runs through the main central shaft of the tank and is controlled from above. Originally, the central cone gate was opened first during flushing and as much sand as possible was run out through this opening by gravity. When sand would no longer run through the central gate by gravity the side gates were opened and the sand was flushed out through the side gates hydraulically by means of a hand-operated fine hose. For some years, however, a mechanical flusher has been in use which rotates on the central shaft of the tank and flushes all sand to the center and out through the center gate. The side gates are no longer used. The present arrangement is a great improvement particularly in minimizing leakage.

The sand in the tank is supported on a canvas filter which is stretched over cocoa matting. Below this cocoa matting is a steel grating of about 5/8-inch mesh, and this steel grating in turn is carried on a steel grid which rests on the tank bottom and permits free circulation of the solvent under the filter. The canvas filter and the cocoa matting are held tightly in place by a rope which is caulked into a wooden groove around the center gate and the outside periphery of the tank. These cocoa mattings have a life of about 14 months, and the canvas lasts about half that time. Occasionally, it may be necessary to give the canvas a bath or wash with dilute hydrochloric acid owing to the clogging of its mesh with calcium and magnesium carbonate from the calcium and magnesium salts in the lake water used in the process. No corrosion is evident and, except for the deposition of carbonates in the cloth and to some extent in the effluent piping, the renewal of pipes either because of corrosion or of clogging is seldom necessary.

The sand after classification is prepared for leaching as follows: After a tank has been filled as full as possible with sand and the feed switched to another tank the first operation is to drain off the excess water. This is done by means of two decanter pipes on opposite sides of the tank provided with wooden plugs for closing after the decanting operation is complete. The surface of the tank even under the best conditions is not entirely level, a requisite for efficient leaching. The leveling is done by means of water jets directed vertically downward against the sand. These jets are carried on a horizontal radial pipe which is rotated about the central axis of the tank, and they make the tank level in a minimum of time. The excess water used in this leveling process is decanted, and the tank is then ready for the leaching cycle.

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This cycle together with sizing of heads and tails, with assays, are shown in Figure 11. The leaching solution as at present constituted will run about 55 grams per liter of NH_3, 40 grams per liter of CO_2, and 25 to 30 grams per liter of copper of which at least 95 per cent should be in the cupric form. Six hundred and fifty cubic meters of this leach is run on to the sand at first as rapidly as possible without overflowing. During this operation the drain or sewer valve is open so that the water in the interstices of the sand within the tank is being displaced downward by the onflowing leach. All percolation is downward. After about 2½ hours, when about 350 cubic meters of leach has been run into the tank, the effluent solution from the sewer valve will begin to show traces of ammonia and copper. The sewer valve is then closed, and the pregnant solutions are run into their proper effluent pipe line. The rate of flow of the ongoing leach is then reduced to a predetermined figure of about 34 cubic meters per hour until the entire 650 cubic meters is exhausted. The first 15 to 30 cubic meters of the effluent solution is run back into the leach storage tanks, and the next 150 cubic meters which contains the bulk of the cuprous copper and of the total copper is drawn off as rich solution for distillation. The control of the process is to a great extent determined at this point by the volume of the "rich" drawn off. Evidently, for a series of cycles the amount of copper withdrawn from the leaching circuit must equal the amount of copper dissolved from the sand. This result is accomplished by varying the volume of the rich solution. If the copper content of the leaching solutions diminishes the volume of rich solution withdrawn is decreased; if the copper content rises the withdrawal is increased.

After all of the leaching solution has been led on to the sand, it is drained down for about an hour and 170 cubic meters of wash is used. This wash may run as high as 75 grams per liter of NH_3 and 50 grams per liter of CO_2, but it must contain not over 5 to 10 grams per liter of copper. This wash has a slight leaching effect, but its main object is to displace the rich copper-bearing leach now in the interstices of the sand. While this wash is going on at the top and after the rich solution has been drawn off from the bottom the valves are changed so that the effluent solution forms the leach for the next cycle. The volume of leach drawn off for the N-1 cycle will be equal to the 650 cubic meters of leach used in cycle N. After 170 cubic meters of wash has been put on the tank there is another period of drainage and 50 cubic meters of a very strong ammonia wash or distillate is used. This wash may contain as much as 130 grams per liter of NH_3, and its chief function is to remove or wash out the copper which is still absorbed in the sand. The use of this strong distillate wash is recent but it is very effective in reducing the assay of the tailings, particularly on certain types of sand that have a greater adsorbtive power for copper than others.

After the distillate wash, about 70 cubic meters of water is run into the tank. It is drained for about 5 hours and is now ready for the steam wash. A special cover is used for this purpose, and steam at practically atmospheric pressure is led on top of the sand at four points while suction is applied below. The steam as it first enters the cold sand is condensed and forms a very effective water wash, carrying out with it practically all of the remaining copper. The adsorbed ammonia still resists this flow of water, but as the entire 1,000 tons of sand becomes heated the ammonia volatilizes and is carried off through the suction pipes to the condenser, so that at the end of the operation there is little ammonia left in the sand independent of the original strength of the solutions.

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While this washing with ammonia distillate, water, and steam is taking place in the ongoing solutions the effluent solutions are manipulated to retain the cycle balance. After the 650 cubic meters of leach, the wash is drawn off. This wash amounts to approximately the same volume as put on. The wash is not drawn off according to volume but according to assay, however, that is, the steaming, and more particularly the suction, is continued until the final effluent wash shows only 1 gram per liter of NH_3 at which point the economical balance between retained ammonia and the use of steam has been reached.

This wash completes the cycle so far as the solution of the copper from the sand is concerned. The ore is now flushed out through the center gate as previously described, and the tank is ready for the next cycle. This complete operation as at present constituted takes 48 to 52 hours, whereas the original cycle took almost twice this long. The time has been reduced chiefly by the use of solutions stronger in copper and ammonia; and the use of this strong solution has been made possible, in turn, by the introduction of the steam wash.

The distillation process is carried out in a separate plant because this plant was available and was near the boiler house. Rich solutions are pumped from the leaching plant to the still house, and the distilled ammonium carbonate liquors are in turn pumped back into the leaching plant and enter the cycle as distillate wash; also, any incoming ammonia required to replace the loss of solvent is ordinarily mixed with this distillate and used in the same manner.

The distillation process consists of preheating the solutions in a specia preheater, distilling or driving off the ammonia and carbon dioxide by means of process steam, partial concentration and cooling of these gases in the dephlegmator or reflux column, final cooling and condensation of the NH_3 and CO_2 as ammonium carbonate, and collecting and filtering the precipitated cuprous and cupric oxide. At present there are three such units, each with a maximum capacity of about 20 cubic meters per hour.

After preheating, the rich solution from the leaching plant goes into the second ring of a 14-section still. These stills were furnished by the Semet-Solvay Co., and while they are similar in principal to the ordinary ammonia still for the distillation of by-product coke-plant liquors they are especially designed with large quick opening doors and large ports and overflows to minimize choking and to make possible maximum speed and accessibility in cleaning. Most of the precipitated oxide remains in suspension throughout its progress through the still and is discharged along with the waste liquors from a trap at the bottom into conical settling tanks. Some of the oxide, however, settles out in the various parts of the stills and, more particularly, bakes on to all of the superheated exposed portions so that after 24 to 48 hours overflows are so seriously clogged that a general cleanout becomes necessary. The quick opening doors provided for this purpose and the general arrangement of the units is such that this cleaning can be done by five men in six hours. It is rather laborious, however, and, as operators appreciated that it would be necessary with a fixed type of still, such frequent cleaning was avoided for many years by using a revolving-shaft type of still especially designed for this process. With the old revolving-shaft stills a general cleanout was required only every four to six weeks instead of every 36 hours as at present, but the advantages of the present system are marked as the stills are simple and compact in structure and efficient mechanically with a

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minimum amount of piping and auxiliary apparatus to clog up. The quick-opening doors are of the same type as those used on the Kelly filter press and were supplied by the Oliver Filter Co.

The suspended oxide after settling out in the cone settler is pumped by means of a Dorrco diaphragm pressure pump to an American filter, and the filtered oxide is dumped directly into cars for transportation to the smelter. This oxide, running over 80 per cent of copper and about 5 per cent of moisture, presents no difficulties in smelting. The overflow of the cone thickeners and the American filter is very dilute and is delivered to a Sweetland filter for continuous filtration.

The steam that enters at the base of the still column passes up through the various sections becoming enriched in its progress with the ammonia taken up from the descending copper ammonia solutions. At the top of the still unit this ammonia vapor may run about 7 per cent of NH_3, which is too dilute for use in the leaching plant or at least at this dilution would unnecessarily increase the volume of leach solutions. Accordingly, this hot vapor is passed to a reflux condenser or dephlegmator where the highest boiling portion (water) is partly condensed and the vapor is concentrated to 15 to 20 per cent of NH_3. The condensate from the reflux is returned to the still. The vapors enriched in ammonia go on to the condenser, are cooled to the liquid state, and are returned to the leaching cycle.

The rich solution as distilled in the still house will run about 40 grams per liter in copper and may vary 5 per cent on either side of this amount in ammonia. The steam used in this operation has been reduced in pressure from 180 pounds at the boiler house by passing it through a 1,250 kw. DeLaval steam turbine exhausting to the stills at 15 to 18 pounds. By this process 500 to 800 kw. are obtained for a minimum expense, and the combination of turbine and still has been found simple and effective. Arrangements are now being made to run the turbine at its full capacity, using such steam as is necessary in distillation and having excess steam go to the large low-pressure turbines which take the exhaust steam from the stamps at about 15 pounds absolute pressure.

Summary of the costs for the last six years is shown below. The table shows a very healthy condition from year to year.

Leaching-plant costs

	1924	1925	1926	1927	1928	1929
Tons treated	1,547,410	1,867,460	1,614,630	1,964,070	1,626,880	1,562,060
Pounds of copper produced	14,054,000	15,221,500	16,194,000	16,153,000	16,757,000	19,599,000
Cost per ton of sand	36.96	29.33	34.73	23.44	23.45	23.02
Recovery, pounds per ton	9.08	8.15	10.03	8.22	10.30	12.55
Cost per pound of copper	4.07	3.60	3.47	2.85	2.34	1.83
Assay of oxide, per cent	82.46	81.56	81.01	81.18	81.53	81.74
Assay feed	.567	.525	.640	.509	.607	.736
Assay tailings	.118	.115	.132	.093	.088	.108
Extraction, per cent	79.2	78.1	79.4	81.7	85.5	85.3

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The decreases in costs shown in 1927 over those in 1926 were the result of a number of intensive investigations and were due to a decided increase in efficiency of distillation resulting from a change in the system, aided by a lowering of the basic cost of ammonia. The further increase in metallurgical efficiency in 1928 over that in 1927 was due chiefly to the introduction of the distillate wash which aided extraction materially. The costs per ton have shown no decrease in 1928 and 1929 over 1927, but the balance has been maintained in the face of a decidedly lessened tonnage and a 10 per cent increase in wages. The costs shown include everything from the delivery of the heads into the leaching plant through delivery of tailings to the tailings dumps and of concentrates to the smelter. They do not include smelting costs. The cost per ton and the cost per pound of copper are probably the lowest in the history of leaching anywhere.

The ammonia loss per ton of sand treated has been running just under one-half pound of NH_3 per ton. The distribution of total cost per ton of sand treated between the major items of operation for the year 1929 is as follows:

General expense	$ 4.28
Conveying and classification	2.72
Leaching	
Ammonia	3.65
Steam	3.89
Other	1.97
Distillation	
Steam	3.44
Other	3.07
Total	$23.02

CONVEYING

After this detailed description of the process of recovering the copper values, one may consider some of the auxiliary apparatus and devices. Because of the wide range of crushing in the stamps, there is not a belt conveyor in the flow sheet. The shaking pan feeds the ore from the rock bins into the mortar hopper, and all conveying thereafter is by means of water in wooden launders. Except for occasional handling of middlings by small centrifugal pumps, the one big lift required between the mill and the regrinding plant for coarse sands, and between the mill and the leaching or flotation plant for fines, is by means of sand wheels.

These warrant a full description. The sand wheels, erected in the early days of the concentrator, were developed by E. D. Leavitt through successive increases in height made necessary by the greater elevation required to reach the receding water line. Before the days of efficient centrifugal sand pumps there was probably some excuse for the higher cost of an elevating device of this type. Even now the mechanical efficiency and operating cost of a 60-foot sand wheel is doubtless less than that of a pump for the same purpose, but the capital expenditure and space required now would be prohibitive. Once installed and paid for, however, the two 50-foot and one 60-foot wheels are dependable, efficient, and all but everlasting.

The sand wheel is a bucket water wheel revolving on a horizontal shaft.

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The buckets open inward to receive the water and tailings when at the bottom and empty when approaching and passing the highest point of their revolution. The wheel has two complete rows of buckets mounted symmetrically on each side of the driving rim. It receives from two lower launders, one on each side, and discharges into two upper apron launders, which converge into one after clearing the wheel. The driving rim carries the driving gear on its outer circumference and is supported from the shaft by tension rods. The buckets are made of flat steel plates. In the 60-foot wheel these plates are set at an angle of 30° to the tangent of the circle drawn through their inner edges. There are 272 buckets on each side with effective widths of 4 feet 3½ inches and radial depths of 20 inches.

The diameter at the inner edge of the buckets is 58 feet 6 inches, at the center of the buckets about 60 feet 2 inches, and at the pitch line of the gear 65 feet. The effective lift, measured from the running surface of the entering launders to the running surface of the discharge launder as it leaves the building, is 49 feet 10 inches. The capacity of the wheel is about 40,000,000 gallons of water and 6,500 tons of sand per 24 hours. Power is supplied by a 700-hp. motor connected by rope drive to a pulley on the shaft of the pinion which meshes with the 65-foot gear.

Each of the 50-foot wheels are driven by a 350-hp. motor belted to the pulley of the pinion shaft. Each side of these wheels has 222 buckets set at 36° to the tangent and a diameter at the inner edge of the buckets of 47 feet 10 inches. The effective lift is 39 feet 7 inches from the running surface of the launders entering the wheel to the corresponding point on the launders leaving the buildings. Each wheel can handle 30,000,000 gallons of water and 3,500 tons of sand per day.

DEWATERING AND HANDLING OF CONCENTRATES

All concentrates go to the smelter only a half a mile away, so that transportation charges are not a factor. The copper is mostly coarse, or at least granular, and drains readily except for the flotation concentrates and oxide from the leaching plant. Coarser concentrates may go to the cars direct from the wet settling bins and drain in the cars or may be dewatered by a drag belt. The average moisture content of all concentrates other than flotation concentrates and oxide is 7 per cent.

Flotation concentrates are thickened in a Dorr thickener and filtered in an Oliver filter delivering concentrates running 14 per cent of moisture directly to 40-ton bottom-dump concentrates cars. Flotation concentrates dump easily and at the smelter are mixed with coarser concentrates for treatment in melting furnaces. This treatment presents no mechanical or metallurgical difficulties when the concentrates are properly mixed and the concentrates require neither drying nor briquetting.

DEWATERING AND DISPOSAL OF TAILINGS

With the plants erected on the shores of a deep lake the disposition of tailings presents no special problem. The concentrator site is also excellent for reclaiming the deposited tailings, and no better site could have been chosen if

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the recovery of these sands had been foreseen at the time of their deposition. No attempt is made either to recover water during concentration or to conserve it, and a problem is created more by the lavish and at times unnecessary quantity of water used than by its scarcity.

WATER SUPPLY

A 60,000,000-gallon triple-expansion reciprocating pump and a 40,000,000-gallon centrifugal pump supply water for concentrating and condensing part of the condensing water being used again in the concentrator. Water is used freely for flushing and conveying sand in launders as the use of much water is cheaper than the use of steeper launders with less water which require more initial elevation. A ratio of 3-parts of water to one of sand is approximately correct for mine rock. On reclaimed tailings the ratio is much lower than this and no accurate data are available for the water-solids ratio of the combined sands.

SAMPLING

No attempt is made to sample mill heads. Intermediate products and tailings are sampled by hand. Final tailings from the leaching plant are sampled from each tank and the sampling of the flotation plant tailings presents no difficulties. Recovery is based upon the assays of smelter product combined with that of concentrator tailings to give a calculated assay of mine rock from which the final extraction is obtained.

RECOVERY OF CONCENTRATES

The recovery of concentrates at present is in excess of 95 per cent, for, even with the smelter loss included the combined recovery of the mill and smelter is that much. Except for overflow losses, as in concentrates and pebble-mill dewaterers, all final tailings come from either leaching or flotation. The overflows are considered a final loss so far as mine rock goes, but they may settle out in the lake and be returned as a reclamation product.

The copper content of the tailings from leaching is chiefly a factor of the fineness of grinding. The sizing assays of this plant plainly show that the best recoveries would be effected by grinding everything through 60 mesh, but other economic conditions determine a somewhat coarser size. The recoveries by flotation on the minus 200 and by leaching on the minus 100 to 200-mesh product are enough alike to indicate that there is no benefit in grinding finer than 60 to 100 mesh for the sake of increasing the percentage of slimes.

Summarized figures for 1929

Total tons of ore treated	840,960
Days operated	290
Hours operated per day	24
Average tonnage per 24 hours running time	2,900
Average sections running	8½
Average tonnage per 24 hours per section	342
Total tons of concentrates	38,918.5
Average tons of concentrates per day	134

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Percentage recovery hand sorting	0.19
Percentage recovery mortar discharge	2.79
Percentage recovery jigs	30.76
Percentage recovery tables	42.06
Percentage recovery fine grinding	5.62
Percentage recovery flotation	9.34
Percentage recovery leaching	9.24
Ratio of concentration	21.6:1
Ratio of water to sand consumption	32:1
Pebble consumption per ton ore, pounds	5.16
Tons per man per day	16.16

Costs for 1929 based on total tonnage milled in cents

Primary grinding (stamps)	19.2
Gravity concentration on stamp product	6.0
Secondary grinding (pebble mills)	8.1
Gravity concentration following pebble mills	1.2
Flotation	3.9
Leaching	10.1
Tailing disposal	1.7
Superintendence, sampling, etc.	3.1
Total	53.3
Steam for stamping (included above)	8.1
Water for concentrator (included above)	1.9

Same costs based on tonnage actually handled in each operation

	Labor	Water and power	Supplies	Total
Primary grinding (stamps)	8.67	7.88	2.65	19.20
Gravity concentration on stamp product	3.10	2.46	.50	6.06
Secondary grinding (pebble mills)	2.84	8.44	3.59	14.87
Gravity concentration following pebble mills	1.15	.82	.15	2.12
Flotation	2.07	2.74	3.82	8.65
Leaching	6.00	2.49	14.51	23.00
Tailing disposal	.64	.91	.14	1.69
Superintendance, sampling, etc.	2.34	.17	.62	3.13

Major unit items of costs entering into total for 1929

Cost of coal per ton delivered to boiler house	$ 4.27
Cost of steam per thousand pounds, cents	28.12
Cost of water per 1,000,000 gallons	2.27
Cost per kw.h., cents	.58
Average wage	3.90

The concentration of Calumet conglomerate is interesting because of the necessary combination of gravity concentration, leaching of sand, and flotation of slimes, it being one of the few ores requiring all of those processes.

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This method of concentration is necessary as values are metallic and are distributed in all sizes from large masses requiring special handling to microscopic ones. The ammonia leaching plant is the only one of its kind, but the process has been adapted to oxidized ores in Rhodesia.

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Screen sizes and assays

Mesh	Steam-stamp product		Hardinge mill plant						Rough jigs		Hutch table	
			Feed		tailings		Conc.		tailings		tailings	
	% wt.	Assay	% wt.	Assay	% wt.	Assay	% wt.	Assay	% wt.	Assay	% wt.	Assay
+10	14.5	5.61	13.0	.89	-	-	-	-	27.1	1.03	-	-
+20	14.4	4.28	30.5	.89	.2	-	.3	45.90	31.9	.90	13.8	1.09
+28	8.3	4.34	18.8	.86	1.3	.53	.4	84.30	15.4	.77	26.0	.78
+35	6.2	3.80	18.2	.76	6.5	.57	1.2	84.10	14.0	.69	32.7	.59
+48	7.2	2.95	8.7	.79	12.0	.59	5.3	56.90	6.3	.69	16.4	.43
+65	6.1	2.23	3.7	.76	14.6	.62	6.1	53.15	3.5	.79	4.7	.38
+100	6.2	1.91	1.6	1.07	22.3	.64	18.8	33.15	1.4	1.20	2.5	.38
+150	3.6	1.91	.5	4.90	5.1	.59	9.3	27.66	.4	5.15	3.9	1.29
+200	5.2	1.53			11.7	.57	31.6	20.90				
-200	28.3	1.09			26.3	.62	27.0	51.75				
Total	100.0	2.95	100.0	.87	100.0	.57	100.0	39.65	100.0	.88	100.0	.70

Mesh	Leaching plant				Flotation plant					
	Feed		tailings		Feed		tailings		Conc.	
	% wt.	Assay	% wt.	Assay	% wt.	Assay	% wt.	Assay	% wt.	Assay
+10	-	-	.3	-						
+20	.6	-	1.4	.29						
+28	1.4	.59	3.0	.20						
+35	4.5	.58	4.7	.16						
+48	14.4	.52	15.2	.14	.5	.14	.6	.12	.4	2.7
+65	18.4	.53	20.3	.11	1.4	.22	1.3	.19	.3	25.0
+100	25.7	.51	26.4	.10	1.5	.40	1.5	.18	2.7	19.1
+150	9.1	.52	8.7	.07	2.6	.45	2.8	.15	1.8	15.6
+200	17.9	.56	14.4	.06	14.0	.52	13.9	.12	13.2	17.4
-200	8.0	1.32	5.6	.11	80.0	.69	79.9	.10	81.6	38.2
Total	100.0	.60	100.0	.11	100.0	.65	100.0	.11	100.0	34.3

I. C. 6364

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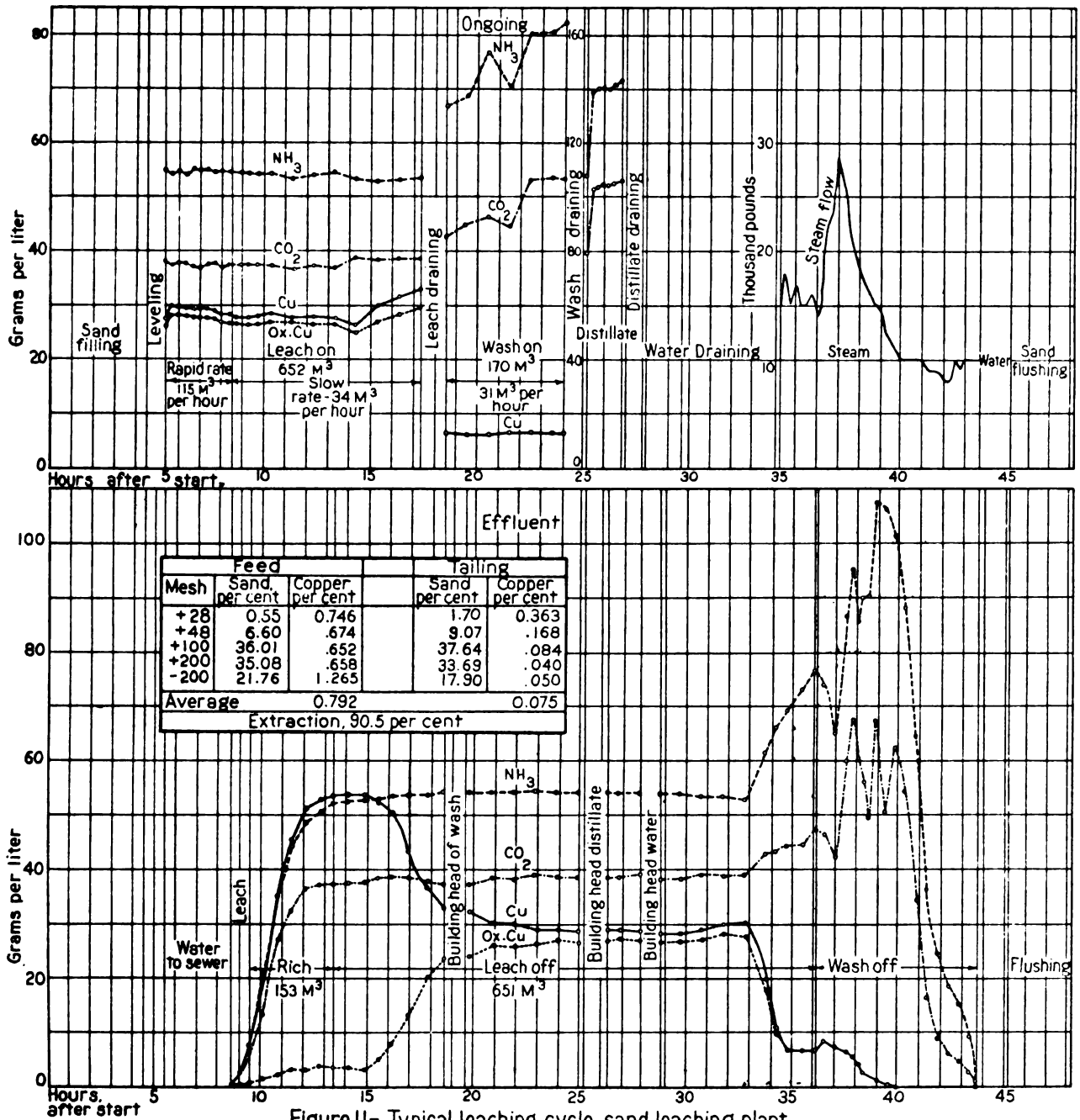

Figure 11.- Typical leaching cycle, sand leaching plant, Calumet and Hecla Mining Co.

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